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MINERAL POLICY SECTOR
INTERNAL REPORT

ALTERNATIVE TECHNOLOGIES FOR METAL PROCESSING

- IMPLICATIONS FOR THE YUKON

D. KING

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Introduction

Lead and zinc concentrates are produced in sufficient quantity in the Yukon to support a separate smelter operation. Cyprus Anvil Mining Corporation, the predominant lead-zinc producer, has expanded its own large ore reserves by acquiring neighbouring properties. Future ore reserves will probably include deposits elsewhere in the Yukon, such as the large deposit at Howard's Pass owned by Placer Development Limited. An agreement is in effect between the Department of Indian and Northern Affairs and Cyprus Anvil, which requires the company to evaluate the feasibility of establishing a smelter operation in the Yukon.

There are regional advantages to be gained from further processing; smelters and refineries are more labour intensive than mine/mills, the added value from concentrates to metals increases foreign exchange; and the establishment of a smelter enables the development of infrastructure and service industries, which may facilitate more developments in the region.

An attempt is made in the following sections to identify the various technical and economic factors that would influence the establishment of such an operation. Projections of economic viability are not included but the types of operations which might be envisaged for the present decade and afterwards are discussed.

This study is confined to lead and zinc, the two most abundant base metals found in the region to date. A smelter producing refined lead would also be able to treat silver bearing concentrates on a custom basis. Copper processing is not considered because up to the present time Yukon copper concentrate production is not sufficient to supply a smelter.

Timing

Two time frames have been chosen for this exercise, namely the current decade, and the years following the 1980s. It is assumed that any plant planned, engineered and installed during the 1980s will be designed on current state-of-the art technology, and will be subject to current regional infrastructure, power and fuel supply and environmental regulations. Plants planned and built during and after the 1990s can incorporate improved technology, including certain processes now at an advanced stage of development, would benefit from any improvements in infrastructure and power supply, but would probably be subject to more stringent environmental regulations.

Ownership

A smelter would probably be corporately integrated with mining and milling operations. This could bring about two different sorts of benefits. Firstly integration would enable the firm to write off its smelter investment against mining profits during smelter construction and thereby significantly improve smelter profitability. For example, the increase in the internal rate of return for an integrated lead smelter versus an unintegrated operation, from a tax point of view, has been estimated at about 2.5 per cent.(1)

Secondly, integration would provide an opportunity to optimize milling and smelter recoveries and costs, which are largely influenced by concentrate grade. The benefit of such optimization would take on added significance if the smelter were to incorporate a process capable of treating bulk or combined lead and zinc concentrates.

The possible gain in overall metal recovery due to the feeding of lower grade concentrates to a smelter would depend on feed composition and process-specific factors. Many ores will yield greater recoveries in producing low grade or bulk concentrates without adding smelting costs to an extent exceeding the value of the extra metals recovered. Alternatively, as in the case of a lead smelter, the optimization of concentrate grade can be geared to the sulphur burning capacity of the facility to maximize its overall utilization of capacity.

Markets

Tariff and non-tariff barriers on metal exports have always presented a major constraint to the domestic reduction of concentrates. It is possible owing to its dependence on foreign energy supplies that Japan will become more amenable to the import of base metals as partial replacement for its present concentrate supplies. Copper and zinc may be cases in point although Japan might be less inclined to import lead metal. The effect of such changes if they occur, would be favourable to the establishment of domestic smelting and to the marketing of the metals from Western Canada.

Labour

Recruitment of comparatively large numbers of skilled personnel is initially more difficult in remote areas than in populated areas. However, over a period of time smelters and refineries tend to provide larger, more stable communities, and attract more service industries than do mines. Cominco Ltd. at Trail and Aluminum Company of Canada, Limited (Alcan) at Kitimat and Arvida are examples of the sort of development and stability that smelters bring to remote areas.

Wage and salary rates for a new facility in the Yukon would no doubt carry a premium or additional fringe benefits. Assumptions can probably be based on rates already existing in the vicinity of Whitehorse or Faro.

Power

The existing developed hydro power is insufficient to accommodate a new smelter complex. Potential hydro sites exist for additional power but its development would depend on a large demand load to produce acceptably low power costs. For example, plans for major hydro development may depend on such potential projects as the electrification of gas pipelines to provide a large enough demand base. Cost of power in remote areas is currently five or six times the cost in lower British Columbia. The annual cost of electricity for a plant would depend on the metal produced and the technology involved in producing it.

Technology in the 1980s

If a green-field plant were to be built in the Yukon in the present decade it would be unlikely to incorporate technology which had not already been commercially proven elsewhere. The commissioning of a plant based on new technology generally involves more problems and expense than conventional plants which are, themselves, not usually free from start-up difficulties. High local costs for labour, equipment and materials would aggravate start-up costs.

The present study assumes that design would be based on best current practice. Even so, two general choices would still be available:

1. treat bulk concentrates for zinc and lead;
2. treat separate commercial grade zinc and lead concentrates in respective installations for each two metals (or one metal only if only one plant is installed).

Treatment of Bulk Concentrates in the 1980s

Treatment of bulk concentrates may be the only option if milling losses become unacceptable in producing separate commercial grade concentrates from other deposits and in any case it may be preferable if an integrated mine/smelter operation is able to optimize mill/smelter recoveries and costs. For the present decade the Imperial Smelting Process (ISP) is the only commercially proven process that can treat bulk concentrates containing substantially more zinc than lead.

In the 1960s many ISP plants were built around the world, nearly all of which are still operating. The popularity of the process appeared to fall off in later years as most zinc plant construction was based on the roast/leach/electrolysis (RLE) process incorporating hot strong acid leaching and iron precipitation by one of the newly patented techniques. The increased price of coke, the cost of meeting environmental standards, and zinc product purity are reasons generally quoted for the ISP's fall from grace. Another reason, seldom mentioned but perhaps decisive for zinc companies with a reliable supply of zinc concentrates, is the reluctance to add lead production and marketing to a predominantly zinc operation which an ISP operation would oblige them to do.

The ISP plant built at Belledune, New Brunswick, was converted to a lead-only blast furnace, apparently because its viability as a lead/zinc operation was impaired by a rise in the price of coke, and because improvements in their milling practice made it more profitable to sell zinc concentrates than produce metal.

A recently published CRU report entitled "Zinc - The Next Decade" concludes that at present the ISP is a more profitable process than electrolysis for the primary production of zinc and that in the future this trend is likely to increase. The cost of energy applied in the CRU report is based on generation of electricity from fossil fuel and therefore some correction may be necessary in applying the conclusions of the report to sites where cheaper power exists, or where the cost of coke, required for the ISP, may be high.

Taking mixed or bulk lead/zinc concentrates as feed, the Imperial Smelting Process essentially comprises sintering followed by a modified lead blast furnace operation in which lead and zinc are reduced to metal, the molten lead passing downwards to the bath and the metallic zinc passing upwards in the form of vapour. Molten lead is recycled to a lead splash chamber in which the zinc vapour dissolves in the lead. The zinc, being less soluble at lower temperatures, later separates from the molten lead when it is cooled. Sulphur dioxide gas from sintering is converted to sulphuric acid for disposal by sale. The literature reveals that a variety of materials and zinc to lead ratios can be tolerated in charges fed to the Imperial Smelting Process, which can also recover copper, silver, gold, cadmium, antimony and bismuth. Recoveries are comparable with separate reduction plants, but overall mill/smelter recoveries can be enhanced by the higher mill recoveries, in producing a bulk concentrate, which can amount to 10 per cent or more.

Coke is required as a reductant for this process and is also the source of about 75 per cent of the energy needed. Electrical power accounts for most of the remaining 25 per cent of energy consumption.

Treatment of Separate Concentrates in the 1980s

The second of the two basic choices, namely the treatment of differentially floated lead and zinc concentrates, has been the more widely applied up to the present time. Commercial smelters, corporately unconnected with the mines, treat concentrates under terms which they negotiate to provide the greatest profitability to the smelter. For example, the smelter would prefer procurement of the purest concentrates possible in order to incur lower operating costs and would set its treatment charges accordingly. The mines also try to negotiate the most favourable terms for themselves, but in the matter of concentrate grade and purity they are faced with several disadvantages. One such disadvantage is that both the zinc roast/leach/electrolysis process and the lead blast furnace process are limited in their operational and economic tolerance of lower grade and impure concentrates. Moreover, smelters are usually located some distance from mines, and low grade concentrates cost more per unit of metal to ship. Competitive factors in the marketing of concentrates have also tended to favour the sale of higher grade concentrates. Up to the present time, therefore, the mines have generally been obliged to produce higher grade concentrates often at considerable sacrifice in mineral recoveries, particularly wherever the minerals are finely dispersed in the ore and are not liberated easily by grinding. The New Brunswick lead zinc ores are an example of ores which yield mineral recoveries of no better than 60-75% in milling to commercial grade concentrates. One of the aims in developing new concentrate treatment technology is to be able to treat low grade and bulk concentrates while maintaining high extraction efficiencies and low treatment costs. Some of the Yukon orebodies being explored are said to be difficult to concentrate into separate commerical lead and zinc concentrates.

About 75 per cent of world zinc is produced in roast/leach/electrolysis plants, in which zinc sulphide concentrates are roasted to produce an oxidic calcine which is leached in sulphuric acid, and the resulting zinc sulphate solution is subjected to electrolysis to extract zinc in the form of cathodes. Sulphur dioxide from the roasting operation is used to produce sulphuric acid. Double absorption acid plants are now generally installed at extra cost to ensure that sulphur dioxide emissions are held to minimal levels, although single-contact plants with neutralisation of tail gas could also meet environmental limits.

Most lead ores are sintered and then fed to blast furnaces to produce a lead bullion which is then refined either in kettles or cast into anodes to be refined electrolytically. The sintering operation does not efficiently utilize the heat of combustion of the sulphide sulphur, and the off-gas is dilute in sulphur dioxide. Coke must be fed to the blast furnace charge in order to reduce the lead oxide sinter to metallic lead. Apart from the low thermal efficiency of the overall process there are environmental problems in the control of sulphur dioxide, dust, and lead fume emissions. While lead plants are meeting current environmental regulations it will become more and more difficult for them to do so as the regulations become more stringent, particularly the older installations.

Technology in the 1990s and Later

Zinc

Developments in zinc extraction technology have followed several broad lines. The 1960s saw the development of the Imperial Smelting Process which was effective in treating mixed or bulk concentrates. In the 1960s and 1970s the efficiency of the roast/leach/electrolysis was greatly improved by incorporation of the additional stages of hot strong acid leaching followed by iron precipitation as jarosite, hematite or goethite.

At the present time the emphasis is mainly on the development of new hydrometallurgical processes to improve efficiences from impure and bulk concentrates and/or eliminate the problem (where it exists) of sulphuric acid by-production, transportation and marketing.

New leaching techniques, which yield an elemental sulphur byproduct, such as the Sherritt Gordon Mines Limited/Cominco Ltd. zinc pressure leaching process, may solve the sulphuric acid problem. The ability to store the sulphur-bearing residues would provide greater flexibility in marketing and remove a potential constraint to metal production, which would prove advantageous in any site location. For a remote location like the Yukon it would also remove the need to transport or otherwise dispose of by-product acid. To some extent this saving would be counterbalanced by the need to bring in fuel or electrical energy to replace the unused heat of combustion of sulphur plus the need to import or produce additional leaching reagents.

Among those processes aimed at treating zinc and bulk concentrates are the Sherritt/Cominco process and the Research and Productivity Council (RPC) sulphating roast process which are outlined in further detail below. Other hydrometallurgical techniques under investigation are based on chloride leaching. There does not appear to be much current interest in developing new pyrometallurgical processes for treating zinc concentrates, although systems have been proposed, eliminating sintering and the use of coke, which should be more energy efficient and should simplify environmental control. These schemes might involve fluid bed roasting and the use of oxygen instead of air with finely divided carbon for the reduction stage.

Sherritt/Cominco Zinc Process

One of the main features of this process is that sulphide sulphur is fixed as elemental sulphur thereby avoiding the problem of sulphuric acid co-production and marketing, and eliminating sulphur dioxide emissions. The elemental sulphur produced can be readily stored or separated from other residues and used or sold.

The essential step in the process is a sulphuric acid pressure leach which eliminates roasting of concentrates. Purification and electrolysis of the pregnant solution are carried out in a manner similar to roast/leach/electrolysis practice. Leach residue treatment, in preparation for lead and silver recovery, constitutes a special stage of the process to achieve the removal of elemental sulphur and pyrite, and possibly a further up-grading to make the residue acceptable as feed to a lead plant. Another special stage may be necessary for the removal of chloride and fluoride ions which, in the conventional roast/leach/electrolysis process, are volatilised during roasting. The leaching stage of the Sherritt process requires an input of sulphuric acid, and since unusable dilute acid is produced during the precipitation of jarosite residue it must be neutralised with limestone. The net consumption of acid must be provided for, either by purchasing it or by burning byproduct elemental sulphur in an on-site acid plant.

A commercial plant is now being installed by Cominco to operate integrally with its existing roast/leach/electrolysis plant. Operating data from this plant should become available over the next few years consequently improving the design definition of future plants. Of particular interest would be data showing how halides can be eliminated from the leach solution, how the grade of the lead residue can be improved and how the recovery of the silver can be increased. This is to be a one-stage leach plant feeding its pregnant liquor into Cominco's existing electrowinning plant, and it will receive low-halide Sullivan concentrates as feed. A greenfield plant may, on the other hand, have to accept the burden of a higher-halide feed and include a step for the removal of chlorine and fluorine. The present commercial installation nevertheless constitutes a major step in the development of this process.

Metal recoveries from various concentrate feeds are reported to be comparable with the conventional roast/leach/electrolysis process although the grade of the lead-silver residue is adversely affected by the amount of iron in the feed concentrate. Special upgrading treatment may be necessary to improve the marketability of the lead-silver residue particularly when bulk concentrates are used as feed.

RPC Sulphation Zinc Roast Process

Where the marketing and shipment of byproduct sulphuric acid is not a problem, a processing alternative such as the RPC sulphation roast can be considered. This process seems to have a good chance for application in northeastern New Brunswick. It has shown promise in small scale pilot tests in treating bulk lead-zinc concentrates and pyritic tailings, and several operating companies have shown interest in its further development.

In this process, zinc or bulk concentrates are roasted at a relatively low temperature in order to convert zinc and copper sulphides to soluble sulphates, and iron sulphide mainly to iron oxide and zinc ferrites. The zinc and copper sulphates are then dissolved in a neutral leach, and any iron in the leach solution is oxidized and precipitated before the pregnant liquor is advanced to a conventional purification system and thence to electrowinning. The leach residue is subjected to a hot strong acid leach, which dissolves the ferrites, and the liquor from this stage is injected into the roaster where the water evaporates and the sulphates decompose to a mixture comprised of hematite, zinc sulphate, and zinc oxide, with evolution of sulphur dioxide and steam. When the solid roaster discharge is recycled back to the neutral leach stage RPC claims that the zinc compounds dissolve completely but that none of the hematite is leached in this stage nor in the subsequent hot strong acid leach. The residue from the hot acid leach contains lead and silver in addition to the hematite. After flotation of the residue, a silver-rich fraction is subjected to a brine leach to remove the lead as lead chloride, which is then treated with lime to produce a lead oxychloride product. Calcination of the lead oxychloride produces calcium plumbate, which, mixed with the silver concentrate, is readily acceptable by a lead smelter.

Although carried out at a relatively low temperature, the sulphating roast stage eliminates any halides present in the concentrate. On the other hand, the off-gases, passing into the cooling and dust collection train to the acid plant, contain some sulphur trioxide and a high percentage of water vapour which may lead to severe corrosion problems upon cooling below the dew point, also to inadequate gas cleaning because of the formation of ferrous sulphate on electrostatic precipitator electrodes. Only an extended large-scale pilot or semi-commercial scale operation would provide reliable operating data for plant definition of the off-gas train. The cost of piloting the RPC process could perhaps be reduced by carrying it out at an existing roast/leach/electrolysis plant.

Lead

There is an urgent need for improvements in lead extraction and refining technology, particularly with regard to environmental regulations and workplace hygiene, but also to improve process economics and energy efficiency. Unlike zinc the main emphasis is on pyrometallurgical processes, and on the direct reduction of concentrates in particular. Less publicized but equally valid is the need to supplant lead fire refining by electro-refining for environmental and hygienic reasons.

Direct reduction of lead concentrates achieves the important effect of eliminating the environmentally troublesome and thermally inefficient sintering stage of conventional processing. However, from the point of view of gas and dust containment, this advantage is lost to some extent in certain of the proposed direct reduction processes in which bullion and or slag must be transferred to and treated in further stages.

Work has been done on a number of different direct reduction processes of which brief mention will be made here of only the Boliden Canada Limited process, the Boliden Top Blown Rotary Converter (TBRC) process, and the Queneau Schuhmann Lurgi (QSL) process, while more detailed comments will be made about the Kivcet CS process. A brief note of one hydrometallurgical lead process, the United States Bureau of Mines (USBM) Chloride Leaching Process, will also be included.

The forerunner of direct reduction processes, the Boliden Process has been commercially applied for many years. However, it involves the transfer and conversion of high sulphur bullion, and more than 40 per cent of the weight of the concentrate reports as dust losses which must be separated from off-gases and recycled. This process has never been adopted by other companies and its environmental disadvantages have led Boliden to adapt the Top Blown Rotary Converter to the smelting of lead concentrate with all stages carried out, batch-wise, in one vessel.

Although no TBRC plant yet exists for the treatment of lead concentrates one is being planned and this must be considered one of the more developed of the new processes. The dusty off-gases from the TBRC will fluctuate in volume and sulphur dioxide strength and will require special care in collection and cleaning.

In the QSL process concentrates are pelletized before being fed to the reactor with powdered coal and oxygen, producing bullion slag and high strength sulphur dixoide. The process has been developed to a large scale batch demonstration stage at Duisburg, West Germany.

Hydrometallurgical lead processes being investigated are invariably based on chloride leaching and include the USBM lead leaching/fused salt electrolysis process.

Kivcet CS Process

The Kivcet CS process originated in the Soviet Union and is licensed for development by Klockner-Humboldt-Deutz. Like several other processes under development it eliminates sintering, is based on the direct reduction of lead sulphide concentrates and has apparently been developed to a large pilot plant scale by the Russians. A commercial scale plant is presently under construction in Bolivia. Cominco is evaluating the Kivcet process for replacement of its sinter/blast furnace capacity, and likewise AMAX Inc. is said to be considering it for replacement of its Missouri lead capacity.

Although considered to be primarily a lead process, one of the advantages claimed for the Kivcet process is its ability to recover zinc either by collecting condensed zinc vapour or by allowing the vapour to reoxidize to zinc oxide dust for collection and marketing.

In this process, sulphide concentrates mixed with pulverized fluxes are blown into a smelting shaft with technically pure oxygen. The material is roasted and smelted while falling into a bath of molten lead rich in metal oxide. The melt continuously enters an adjoining electric furnace by passing below a water-cooled partition separating the shaft from the electric furnace.

The process dispenses with the need for a sintering stage, thus avoiding the recycling of feed materials and reducing the dilution of sulphurous off-gas. Since pure oxygen is used, the SO_2 off-gas is of high strength, volumes are minimal and dust collection systems are correspondingly small. Coke fines are used as the reductant in the electrothermal section of the furnace, but it should also be possible to substitute coal for some of the coke reductant.

USBM Leaching Process

This process, which has been developed to a large scale continuous pilot stage producing 500 pounds of lead per day, involves ferric chloride leaching of lead sulphide concentrates and recovery of lead by fused salt electrolysis of lead chloride to produce lead metal and chlorine. The chlorine is recycled to regenerate ferric chloride for leaching. Sulphide sulphur is converted to elemental sulphur which passes out of the system with the leach residue.

The USBM reports good success in its tests with no major operational problems. The corrosive chloride media is reportedly accommodated satisfactorily by various plastic materials and by titanium.

Larger scale and possible continuous testing may be required to define solid and liquid waste disposal and examine the build-up and control of impurities in the leach solution and electrolyte. So far, USBM has observed no consistency in the build-up of sulphate, sodium, iron, copper and zinc.

Lead leaching recovery is reported to be about 98 per cent, with silver extraction about 80 per cent, copper 80 per cent, and zinc 60 to 70 per cent. It has not been made clear how copper, zinc and silver would be recovered from the leach solution.

Quantitative Review

Process Combinations

The choice of process for a given location would be largely governed by the availability and cost of input energy and supplies to the plant, and also by the types and quantities of products and byproducts.

Each type of process has its own characteristic combination of plant inputs and outputs. In this study a quantitative review is made of only five processes representing the basic scenarios outlined in the foregoing sections.

The case of bulk concentrate treatment in the 1980s is considered on the basis of the Imperial Smelting Process while the treatment of separate concentrates is examined on the basis of a combination of the roast/leach/electrolysis for zinc and the sinter/blast furnace process for lead. Each of these processes produces byproduct sulphuric acid. For the 1990s and beyond, the two processes selected for quantitative review are the Sherritt/Cominco pressure leach process for zinc concentrates, and the Kivcet CS process for the treatment of lead concentrates. Assuming the successful resolution of all developmental problems for these two processes there appears to be certain potential advantages in operating each in collaboration with the other. For one thing it may be possible to make these two processes complement each other to some degree with regard to their sulphurous outputs. To what extent this may be possible will depend on a resolution of the question of acid consumption by the Sherritt/Cominco process for two-stage leaching and possibly of the halide removal and lead residue upgrading questions. The acid consumption estimated in this study is based on currently available information and, as such, it is subject to modification as the procedures for halide removal and lead residue upgrading becomes finalized.

Another benefit to be gained by matching these new processes would be increased metal recoveries, each process helping the other. The lead/silver residue from a Sherritt/Cominco zinc plant could be treated in an adjacent Kivcet CS plant, thereby avoiding transportation costs on the residue. Furthermore, the Sherritt/Cominco zinc process can, if necessary, accept bulk zinc/lead concentrates, while the Kivcet CS lead process can treat lead concentrates high in zinc and recover the zinc as impure zinc oxide for recycling back to the zinc plant. Savings due to shared infrastructure labor and overheads would of course apply to any combination of plants, but except for the ISP, the older processes cannot accept bulk concentrates and zinc cannot be recovered from the conventional lead blast furnace operations unless an additional slag fuming plant is installed.

Environmentally the use of a process such as the Kivcet CS process with electro-refining of the lead bullion would be a considerable improvement over the sinter/blast furnace with fire refining. The use of tonnage oxygen in the process minimizes the volume of off-gases which are correspondingly stronger in sulphur dioxide. Because of the small off-gas volume the containment of lead fume and dust should prove easier than in current commercial practice leading to lower emissions. The Sherritt/Cominco zinc pressure leaching process requires no roasting stage and fixes sulphur in elemental form enabling flexibility and control over the quantity of gypsum to be discarded, as well as eliminating the need for off-gas cooling and dust collection.

There would appear be little advantage in matching a chloride leach lead process with any sulphate-based zinc process because there would be no opportunity for either process to supply the other with reagent. However should the lead chloride leach prove far superior from an environmental viewpoint this factor might prove to be overriding.

Inputs and Outputs

Calculations of material flow are based on zinc and lead production of 100 000 and 50 000 tonnes per year respectively, assumed to be roughly the ratio of the two metals in the Yukon ores.

The feed concentrates are assumed to have the compositions shown in Table 1, in which the blended feed for the ISP represents the mixture of zinc and lead concentrates which would yield the required outputs of zinc and lead metals.

Recoveries of zinc, lead and fixed sulphur are assumed from published data and from informal sources to be as shown in Table 2 for each of the processes discussed. These assumptions are based on published data and on information provided by the designers and licensors of new processes.

Calculated tonnage flows of plant materials are listed in Tables 3, 4 and 5 together with electrical power consumptions in kilowatt hours. The quantities for the paired zinc and lead plants are combined to provide totals of inputs and outputs for each plant complex. In Table 3 a breakdown of quantities is provided for a plant complex comprising a roast/leach/electrolysis zinc plant and an adjacent sinter/blast furnace lead plant. Table 4 provides a breakdown for an Imperial Smelting Process to produce both zinc and lead, while Table 5 indicates the input and output quantities one might expect from a side-by-side combination of a Sherritt/Cominco zinc plant and a Kivcet lead plant.

Power consumption for the Imperial Smelting Process is only 91 million kilowatt hours unlike the other two plant complexes which each incorporates zinc electrowinning and uses about 450 million kilowatt hours annually. On the other hand the ISP uses about 95 000 tonnes per year of coke and 194 000 tonnes per year of fuel oil, compared with total coke, coal, and oil consumptions of about 25 000 tonnes per year for the RLE-S/BF complex and 34 000 tonnes per year for the S/C - Kivcet complex.

Concentrate inputs to the RLE-S/BF complex are significantly greater than to either of the other plants because it lacks the facility to recover zinc from the lead concentrates. The differences amount to 1.3% and 2.6% more lead concentrate, and 3% and 4% more zinc concentrate than for the ISP and S/C - Kivcet plants respectively.

TABLE 1

Feed Compositions Assumed in Estimating Plant Inputs and Outputs

	Zinc Concentrate (%)	ISP Blended Feed (%)	Lead Concentrate (%)
Zinc	52.0	38.8	7.0
Lead	1.5	20.4	66.0
Iron	8.0	6.83	4.0
Sulphur	32.0	27.3	16.0

TABLE 2

Processing Recoveries Assumed in Estimating Plant Inputs and Outputs

Process	Overall Recoveries - %		
	Zinc	Lead	Sulphur as H_2SO_4
RLE (zinc)	96	85	85
S/BF (lead)	0	95	95
ISP (zinc/lead)	94	95	97
S/C (zinc)	96	85	85 (as Elemental S)
Kivcet CS (lead)	78	97	97

TABLE 3

Material Flow (1980)
Option No. 1 - RLE Zinc with S/BF Lead Plant

Output	RLE	S/BF	Total
		(tonnes/year)	
Lead	2 426	47 574	50 000
Zinc	100 000	0	100 000
Sulphuric Acid	166 867	36 329	203 196
Copper Dross		4 000	4 100
Slag		32 000	32 000
Lead/Silver Residue	9 227		9 227
Jarosite	49 712		49 712
Inputs			
Lead concentrate		75 876	75 876
Zinc concentrate	200 320		200 320
Sulphuric acid	8 000		8 000
Lime	1 600	200	1 800
NH_4Cl	80		80
MnO_2	200		200
Sb_2O_3	300		300
Limestone		8 000	8 000
Iron ore		16 000	16 000
Silica		4 000	4 000
Soda Ash		800	800
Sulphur		500	500
H_2SiF_6		100	100
Coke		15 500	15 500
Fuel oil	3 300	6 600	9 900
Electricity	433 x 10 kwh	17 x 10^6 kwh	450x10^6 kwh

TABLE 4

Material Flow (1980)
Option No. 2 - Imperial Smelting Process for Mixed Concentrate

	Tonnes per year
Outputs	
Lead	50 000
Zinc	100 000
Sulphuric Acid	220 649
Copper Dross	4 100
Slag	65 000
Inputs	
Lead concentrate	75 326
Zinc concentrate	194 441
Coke	95 500
Fuel oil	194 441
Lime	200
Limestone	21 000
Silica	3 000
Soda ash	800
Sulphur	500
H_2SiF_6	100
Electricity	91 x 10^6 kwh

TABLE 5

Material Flow (1990s and Later)
Sherritt/Cominco Zinc with Kivcet CS Lead Plant

	S/C Zinc	Kivcet Lead	Total
		(tonnes/year)	
Output			
Lead	2 450	47 550	50 000
Zinc	95 945	4 055	100 000
Sulphuric Acid		35 303	35 303
Sulphur	52 000		52 000
Zinc Oxide Dust		5 053	
Copper Dross		4 000	4 000
Slag		20 000	20 000
Gypsum	51 000		51 000
Lead Jarosite	15 500		15 500
Iron Residue	20 450		20 450
Steam		70 000	70 000
Inputs			
Lead concentrate		74 273	74 273
Zinc concentrate	192 196		192 196
Sulphuric acid	34 500		34 500
Oxygen	90 000	65 000	155 000
Steam	200 000		200 000
Limestone	30 000	4 000	34 000
Lime	1 600	240	1 840
Flocculant	9 000		9 000
Iron ore		3 500	3 500
Sand		6 000	6 000
Soda ash		800	800
Sulphur		500	500
Fluosilicic acid		100	100
Coke		5 000	5 000
Fuel oil			700
Coal (for steam)	29 000		29 000
Electricity	400×10^6 kwh	58×10^6 kwh	458×10^6 kwh

However, the most notable difference in material flow exists with regard to the by-produced sulphur products. Both the RLE-S/BF and the ISP produce over 200 000 tonnes of sulphuric acid per year, which would have to be shipped out. In contrast the S/C - Kivcet produces about 50 000 tonnes of elemental sulphur through the S/C zinc plant, for local storage or disposal, and about 35 000 tonnes per year of sulphuric acid through the Kivcet lead plant of which 34 000 tonnes would be consumed in the S/C zinc plant. This near balance in sulphuric acid production and consumption coupled with the production of storable elemental sulphur would virtually eliminate the serious problem of sulphuric acid production and marketing.

Transportation of Plant Materials

The impact on transportation requirements of the different processes and process combinations is explored, in terms of tonne-miles per year, using two possible alternative plant locations, namely Whitehorse and the vicinity of Faro and Pelly River Terminal.

Depletion of ores in the Faro district coinciding with the development of tonnage in other areas will probably bring about future changes in the mileage for hauling concentrates to the smelter. It is assumed that the new supplies would eventually come from farther north than Faro. Two cases are therefore considered for the source of concentrate supply, namely 100% from the Faro district and 50% from the Faro district with the remaining 50% being trucked from the Howard's Pass area.

Approximate distances between points in the Yukon are listed in Table 6. Annual tonnes, and tonne-miles calculated for each plant material are shown in detail in Tables 7 through 10. It is assumed that slag, tailings and local limestone supplies would be transported a nominal ten miles.

From the point of view of the movement of concentrates the advantage of having a smelter at Faro, compared with Whitehorse, is partially lost when 50% of the concentrate feed is brought in from Howard's Pass. The concentrate tonne-miles per year increases from 2 761 to 25 549 while in the case of a Whitehorse smelter the concentrate tonne-miles per year increases from 55 792 to 74 850. On the other hand, the advantage of a Whitehorse site, with regard to the outward transportation of large quantities of sulphuric acid (as from the RLE-S/BF and ISP plants), is unaffected by the change in the source of concentrates; the annual sulphuric acid tonne-miles from Whitehorse remains at 23 000 to 24 000 tonnes compared with 63 000 to 68 000 from a Faro district smelter.

In the case of the S/C - Kivcet complex, which disposes of its elemental sulphur residue and internally consumes nearly all of its by-product sulphuric acid, a Whitehorse location has no such advantage over Faro. However, a Whitehorse location is favoured with regard to all plant supplies which must be imported into the Yukon (notably coke in the case of the ISP), and all metal bearing products which must be sold outside of the Yukon.

TABLE 6

Yukon Road and Rail Distances

From	To	Miles	
Faro/Pelly River Term	Whitehorse via Ross River	237	(Road)
" " "	Whitehorse via Carmacks	202	"
" " "	Jason/Tom deposits	174	"
" " "	Summit/Howards Pass	178	"
" " "	Keno Hill	225	"
" " "	Carmacks	96	"
" " "	Ross River	43	"
" " "	Albert/Watson Lake	256	"
Ross River	Tom	126	"
" "	Assumed turn-off to Howards Pass	65	"
Howards Pass turn-off	Howards Pass	70	"
Whitehorse	Tom	335	"
"	Summitt/Howards Pass	344	"
"	Junction near Ross River	209	"
"	Johnson Crossing	83	"
Johnson Crossing	Albert/Watson Lake	174	"
Whitehorse	Albert/Watson Lake	257	"
"	Carmacks	106	"
Carmacks	Keno Hill	176	"
Whitehorse	Keno Hill	282	"
Whitehorse	Skagway	110	(Rail)

TABLE 7

Transportation of Input/Output Materials (Annual Tonne-Miles)
(Feed Concentrates: 50% Faro - 50% Howard's Pass)
(smelter at Faro/Pelly River)

Material to be Transported	Miles	RLE Zinc + S/BF Lead		ISP Zinc/Lead		S/C Zinc + Kivcet Lead	
		tonnes per year	tonne-miles per year$\times 10^3$	tonnes per year	tonne-miles per year$\times 10^3$	tonnes per year	tonne-miles per year$\times 10^3$
Outputs							
Lead	312	50 000	15 600	50 000	15 600	50 000	15 600
Lead/silver residue	Recycled	9 230	0	0	0	15 500	0
Zinc	312	100 000	31 200	100 000	31 200	100 000	31 200
Zinc oxide dust	Recycled	0	0	0	0	5 053	0
Copper Dross	312	4 100	1 279	4 100	1 279	4 000	1 248
Slag	10	32 000	320	65 000	650	20 000	200
Leach residue	10	49 700	497	0	0	71 450	714
Sulphuric acid	312	203 196	63 397	220 649	68 842	35 300	250
Sulphur	10	0	0	0	0	52 000	520
Total outputs		438 996	112 293	349 749	117 571	352 300	49 732
Inputs							
Lead concentrate	92.5	75 876	7 019	75 330	6 968	74 273	6 870
Zinc concentrate	92.5	200 320	18 530	194 440	17 986	192 196	17 778
Limestone	10	8 000	80	21 000	210	34 000	340
Iron ore	312	16 000	4 992	0	0	3 500	1 092
Silica	312	4 000	1 248	3 000	936	6 000	1 872
Sulphuric acid	Recycled	8 000	0	0	0	34 500	0
Reagents	312	3 780	1 179	1 600	499	12 820	4 000
Coke	312	15 500	4 836	95 500	29 796	5 000	1 560
Coal	312	0	0	0	0	29 000	9 048
Fuel oil	312	9 900	3 089	19 400	6 053	700	218
Total inputs		321 796	40 973	412 270	62 448	391 989	42 778
GRAND TOTAL		770 792	153 266	762 019	180 019	744 289	92 510

Transportation of Input/Output Materials (Annual Tonne-Miles)
(Feed Concentrates: 50% Faro - 50% Howard's Pass)
(smelter at Whitehorse)

		RLE Zinc + S/BF Lead		ISP Zinc/Lead		S/C Zinc + Kivcet Lead	
Material to be Transported	Miles	tonnes per year	tonne-miles per yearx10^3	tonnes per year	tonne-miles per yearx10^3	tonnes per year	tonne-miles per yearx10^3
Outputs							
Lead	110	50 000	5 500	50 000	5 500	50 000	5 500
Lead/silver residue	Recycled	9 230	0	0	0	15 500	0
Zinc	110	100 000	11 000	100 000	11 000	100 000	11 000
Zinc oxide dust	Recycled	0	0	0	0	4 050	0
Copper Dross	110	4 100	451	4 100	451	4 000	451
Slag	10	32 000	320	65 000	650	20 000	200
Leach residue	10	49 700	497	0	0	71 450	714
Sulphuric acid	110	203 196	22 352	220 649	24 271	35 300	88
Sulphur	10	0	0	0	0	52 000	520
Total outputs		438 996	40 120	349 749	41 872	352 300	18 385
Inputs							
Lead concentrate	271	75 876	20 563	75 330	20 414	74 273	20 128
Zinc concentrate	271	200 320	54 287	194 440	52 693	192 196	52 085
Limestone	10	8 000	80	21 000	210	34 000	340
Iron ore	110	16 000	1 760	0	0	3 500	385
Silica	110	4 000	440	3 000	330	6 000	660
Sulphuric acid	Recycled	8 000	0	0	0	34 500	0
Reagents	110	3 780	416	1 600	176	12 820	1 410
Coke	110	15 500	1 705	95 500	10 505	5 000	550
Coal	110	0	0	0	0	29 000	3 190
Fuel oil	110	9 900	1 089	19 400	2 134	700	77
		321 796	80 340	412 270	86 462	391 989	78 825
		770 792	120 460	762 019	128 334	744 289	97 210

TABLE 9

Transportation of Input/Output Materials (Annual Tonne-Miles)
(Feed Concentrates: 100% Faro)
(smelter at Faro/Pelly River)

Material to be Transported	Miles	RLE Zinc + S/BF Lead		ISP Zinc/Lead		S/C Zinc + Kivcet Lead	
		tonnes per year	tonne-miles per yearx10^3	tonnes per year	tonne-miles per yearx10^3	tonnes per year	tonne-miles per yearx10^3
Outputs							
Lead	312	50 000	15 600	50 000	15 600	50 000	15 600
Lead/silver residue	Recycled	9 230	0	0	0	15 500	0
Zinc	312	100 000	31 200	100 000	31 200	100 000	31 200
Zinc oxide dust	Recycled	0	0	0	0	4 050	0
Copper Dross	312	4 100	1 279	4 100	1 279	4 000	1 248
Slag	10	32 000	320	65 000	650	20 000	200
Leach residue	10	49 700	497	0	0	71 450	714
Sulphuric acid	312	203 196	63 397	220 649	68 842	35 300	250
Sulphur	10	0	0	0	0	52 000	520
Total outputs		438 996	112 293	349 749	117 571	352 300	49 732
Inputs							
Lead concentrate	10	75 876	758	75 330	753	74 273	743
Zinc concentrate	10	200 320	2 003	194 440	1 944	192 196	1 922
Limestone	10	8 000	80	21 000	210	34 000	340
Iron ore	312	16 000	4 992	0	0	3 500	1 092
Silica	312	4 000	1 248	3 000	936	6 000	1 872
Sulphuric acid	Recycled	8 000	0	0	0	34 500	0
Reagents	312	3 780	1 179	1 600	499	12 820	4 000
Coke	312	15 500	4 836	95 500	27 796	5 000	1 560
Coal	312	0	0	0	0	29 000	9 048
Fuel oil	312	9 900	3 089	19 400	6 053	700	218
Total inputs		321 796	18 185	412 270	38 191	391 989	20 795
Total Material Flow		770 792	130 478	762 019	155 762	744 289	70 527

TABLE 10

Transportation of Input/Output Materials (Annual Tonne-Miles)
(Feed Concentrates: 100% à Faro)
(smelter at Whitehorse)

Material to be Transported	Miles	RLE Zinc + S/BF Lead		ISP Zinc/Lead		S/C Zinc + Kivcet Lead	
		tonnes per year	tonne-miles per yearx10^3	tonnes per year	tonne-miles per yearx10^3	tonnes per year	tonne-miles per yearx10^3
Outputs							
Lead	110	50 000	5 500	50 000	5 500	50 000	5 500
Lead/silver residue	Recycled	9 230	0	0	0	15 500	0
Zinc	110	100 000	11 000	100 000	11 000	100 000	11 000
Zinc oxide dust	Recycled	0	0	0	0	4 050	0
Copper Dross	110	4 100	451	4 100	451	4 000	451
Slag	10	32 000	320	65 000	650	20 000	200
Leach residue	10	49 700	497	0	0	71 450	714
Sulphuric acid	110	203 196	23 352	220 649	24 271	35 300	88
Sulphur	10	0	0	0	0	52 000	520
Total outputs		438 996	40 120	349 749	41 872	352 300	18 385
Inputs							
Lead concentrate	202	75 876	15 327	75 330	15 217	74 273	15 003
Zinc concentrate	202	200 320	40 465	194 440	39 277	192 196	38 824
Limestone	10	8 000	80	21 000	210	34 000	340
Iron ore	110	16 000	1 760	0	0	3 500	385
Silica	110	4 000	440	3 000	330	6 000	660
Sulphuric acid	Recycled	8 000	0	0	0	34 500	0
Reagents	110	3 780	416	1 600	176	12 820	1 410
Coke	110	15 500	1 705	95 500	10 505	5 000	550
Coal	110	0	0	0	0	29 000	3 190
Fuel oil	110	9 900	1 089	19 400	2 134	700	77
Total inputs		321 796	61 282	412 270	67 849	391 989	60 439
Total Material Flow		770 792	101 402	762 019	109 721	744 289	78 824

In order to obtain overall visibility of transportation data all cases have been condensed in Tables 11 through 13 to show only the totals of input and output tonne-miles per year for each process with respect to smelter location and feed source.

From the condensed tables it is readily apparent that in both the cases of an RLE-S/BF complex and an ISP plant, the overall number of tonne-miles per year required to transport ingoing and outgoing plant materials is considerably lower for a Whitehorse plant site than one in the vicinity of Faro. Moreover, this advantage to Whitehorse becomes more pronounced as more feed concentrate is brought in from more northerly deposits like Howard's Pass.

A "balanced" plant complex based upon a Sherritt/Cominco zinc process and a Kivcet CS lead process would require 20% to 30% less tonne-miles per year than even a Whitehorse plant site. In this case, however, the Faro plant site is somewhat favoured over the Whitehorse location if all other factors such as labour and power availability are equal.

Power Consumption

As mentioned earlier the somewhat greater transportation load of the ISP than the RLE-S/BF, is partly a reflection of the need to import coke for zinc as well as lead reduction. However, the ISP is the only one of the three options which does not require electrical power for zinc electrowinning.

Consequently, in view of the present lack of developed power in the Yukon, the ISP is the only process which could be installed in the 1980s, provided one could overcome other serious constraints such as sulphuric acid marketing and transportation.

Construction of a hydro facility of sufficient size to provide power at a suitably low price for a world scale zinc electrowinning plant, is assumed to require about 12 years, whereas a smelter would only require 3 to 4 years to design and construct. Therefore, suitable developed power could conceivably be available in the 1990s (when it is assumed that the S/C + Kivcet CS would be sufficiently developed), but only if a decision to install power were made in the 1980s.

To plan twelve years ahead in order to provide power supply for projects which require a design/construction period of only 3 to 4 years, would require unusually effective co-ordination and foresight between private industry, hydro authorities and government. From the point of view of power availability, the prospect of installing a smelter complex in the Yukon therefore appears unlikely even in the 1990s, unless hydro power development were to be coupled to projects with large demand such as electrification of railways or electrified pumping and heating for oil pipe lines. Hydro electric and other forms of energy are exportable in the form

TABLE 11

Transportation of Input/Output Materials - RLE Zinc + S/BF Lead Plants

Feed Source and Smelter Location	Tonnes - Miles/Year x 1000		
	Inputs	Outputs	Totals
Feed 100% from Faro Area			
Smelter at Faro/Pelly River	18 185	112 293	130 478
Smelter at Whitehorse	61 394	40 120	101 514
Feed 50% from Faro Area plus 50% from Howard's Pass			
Smelter at Faro/Pelly River	40 973	112 293	153 266
Smelter at Whitehorse	80 340	40 120	120 460

TABLE 12

Transportation of Input/Output Materials - Imperial Smelting Zinc/Lead Plant

Feed Source and Smelter Location	Tonnes - Miles/Year x 1000		
	Inputs	Outputs	Totals
Feed 100% from Faro Area			
Smelter at Faro/Pelly River	38 191	117 571	155 762
Smelter at Whitehorse	67 848	41 872	109 720
Feed 50% from Faro Area plus 50% from Howard's Pass			
Smelter at Faro/Pelly River	62 448	117 571	180 019
Smelter at Whitehorse	86 462	41 872	128 334

TABLE 13

Transportation of Input/Output Materials - Sherritt/Cominco Zinc + Kivcet Lead Plant

Feed Source and Smelter Location	Tonnes - Miles/Year x 1000		
	Inputs	Outputs	Totals
Feed 100% from Faro Area			
Smelter at Faro/Pelly River	20 795	49 732	70 527
Smelter at Whitehorse	60 438	18 385	78 823
Feed 50% from Faro Area plus 50% from Howard's Pass			
Smelter at Faro/Pelly River	42 778	49 732	92 510
Smelter at Whitehorse	78 825	18 385	97 210

of metals which require a large energy input for their extraction from ores. Therefore hydro power can be regarded as an exploitable renewable resource to be developed and sold at a profit, and hydro authorities could be in the forefront of promoting primary metal production as a means of exporting the "locked-in" energy. At present the role adopted by hydro authorities appears to be strictly passive in providing a service upon the initiative of potential consumers.

Conclusions

While it becomes apparent, from the present exercise that prior development of competitively priced electrical power is the key to the development of the Yukon it is also evident that other economic factors will weigh against the establishment of primary metal reduction plants as technology now stands. For example, since transportation clearly favours a Whitehorse rather than a minesite location using current technology, it follows that any Yukon location would be at a competitive disadvantage with a coastal site, even if all other economic factors were equal. It is encouraging to note, on the other hand, that emerging zinc and lead technology can be so "married" as to make a minesite location somewhat more favourable than one closer to the markets. Whether or not this advantage would overcome other commercial factors such as cost of labour, cost of construction, etc. remains for future and perhaps more specific study.

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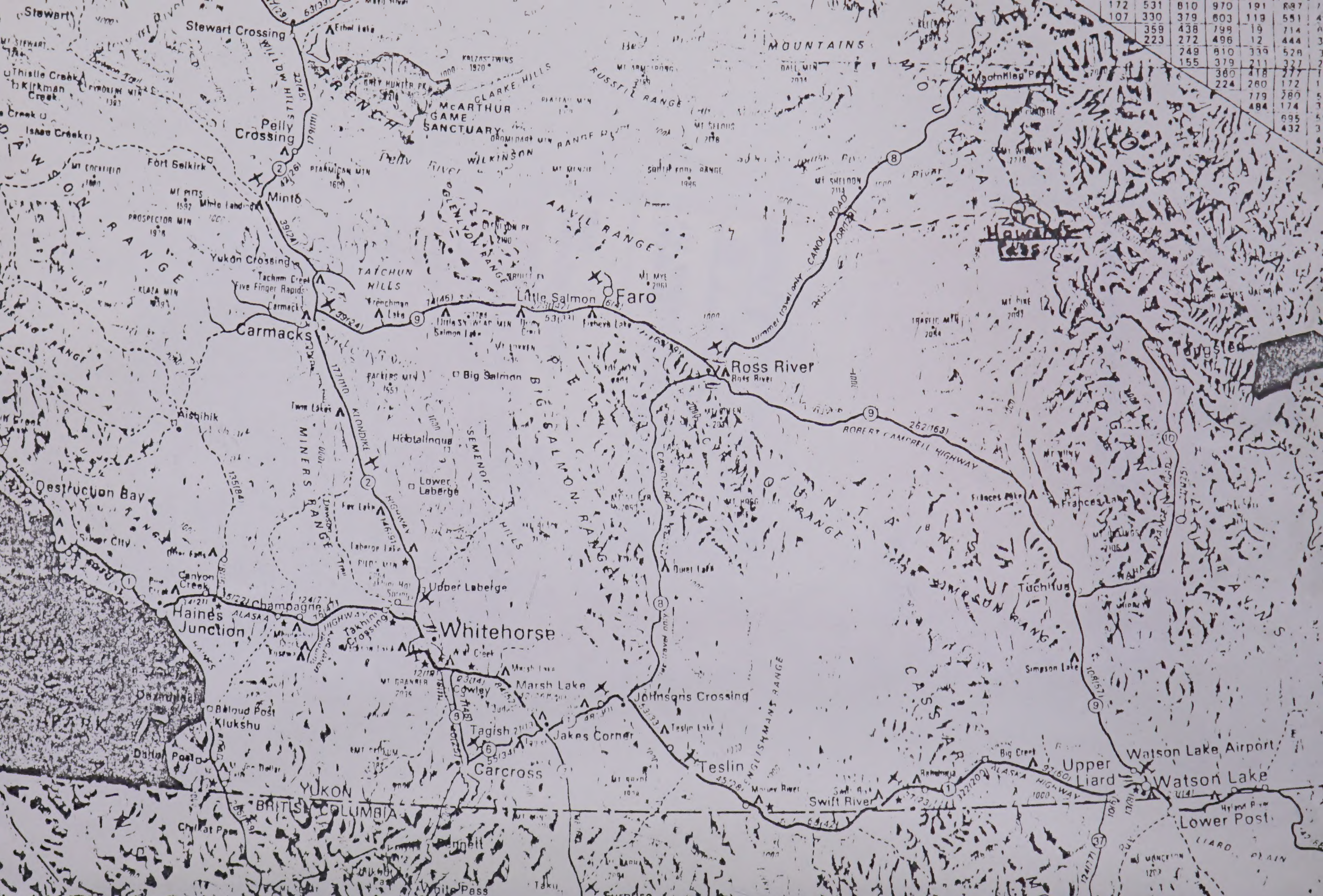
Stewart
Stewart Crossing
Thistle Creek
Kirkman Creek
Isaac Creek
Pelly Crossing
Fort Selkirk
Minto
Minto Landing
Prospector Mtn
Yukon Crossing
Tatchun Creek
Five Finger Rapids
Carmacks
Tatchun Hills
Frenchman Lake
Little Salmon
Faro
Ross River
Fisheye Lake
Big Salmon
Aishihik
Destruction Bay
Silver City
Canyon Creek
Haines Junction
Champagne
Takhini Crossing
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Englishmans Range
Simpson Range
Cassiar Mountains
Dawson Range
Tungsten
Mt Mye 2061
Mt Sheldon 2114
Macmillan Pass
Grey Hunter Pk
Mt Granger 2015
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